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METHODS OF MINING DISSEMINATED LEAD ORE AT A MINE IN THE SOUTHEAST MISSOURI DISTRICT

BY

C. F. JACKSON

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METHODS OF MINING DISSEMINATED LEAD ORE AT A MINE IN THE SOUTHEAST MISSOURI DISTRICT[1]

By C. F. Jackson[2]

INTRODUCTION

This paper describing the mining practices at a mine in the Southeast Missouri district is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States.

The conditions, operating methods, and practice described in the following pages are those obtaining at a porperty in the Southeastern Missouri lead belt, St. Francois County, Mo. This property is the oldest in the county and is a consolidation of a number of separate mines originally opened up as individual operations many years ago. Present-day operations have necessarily to be adapted to suit the older levels, stopes, and pillars, and also the recovery of ore from worked over areas which was considered too low grade to mine in the earlier days constitutes a different problem than that of development and mining in virgin ground.

Quite naturally, since this is the oldest property in the district, many of the methods and details of practice now well standardized for the district have been worked out here, although conditions are not as favorable for their efficient application and for as good results as are the conditions at some of the newer mines, where better opportunity is afforded to plan work to take advantageof them. In this paper an attempt is made to outline the conditions at this particular property and the operating methods and practice employed.

ACKNOWLEDGMENTS

The author acknowledges the many courtesies extended by the manager and operating officials and their assistance and suggestions in gathering and compiling the material for this paper.

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1 - The Bureau of Mines will welcome reprinting of this paper, probided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines, Information Circular 6170."

2 - Mining engineer, U. S. Bureau of Mines.

The historical items have been taken from Geology of the Disseminated Lead Deposits of St. Francois and Washington Counties, Volume 9, parts 1 and 2, Missouri Bureau of Geology and Mines, by H. A. Buehler and E. R. Buckley, and the same source of information has been drawn upon in preparing the paragraphs on geology and ore deposits.

HISTORY

The first mining in St. Francois County, Mo., dates back to the period from 1742 to 1762. Winslow[3] estimated that 1,000 tons of lead ore (500 tons of lead) were produced in the county during the 18th century. This ore was obtained from shallow surface workings above ground water level, the galena occurring as masses and crystal aggregates in caves and crevices.

Disseminated ore was discovered at Bonne Terre in 1864, and in 1869 the St. Joseph Lead Co. introduced the first diamond drill into the district for exploring at depth. This is probably one of the earliest if not the earliest use of the diamond drill in prospecting for ore.

The production of St. Francois County up to 1869 was estimated by Winslow at 124,930 short tons of ore, valued at $5,082,700. From 1869 to 1907 inclusive 1,202,607 short tons of lead concentrates were produced, valued at $59,869,354.

At present the mines of the Southeastern Missouri lead district are producing approximately 500,000 tons of crude ore per month.

The mining method employed is that of open stopes with pillar support, and practically no timber is used. Methods and practices now in use have been developed to suit the conditions and are the result of the experience of about 60 years.

GEOLOGY AND ORE DEPOSITS

The disseminated lead ores of southeastern Missouri occur at various horizons in the Bonneterre dolomitic limestone of Cambrian age. This formation has a normal thickness of about 365 feet and, at the property discussed in this paper, outcrops at the surface in places and comes close to surface over most of the area. The beds lie in an approximately horizontal position considering the area as a whole, though local dips of a few degrees up to a maximum of 20° have been observed and the whole formation has a slight dip.

The Bonneterre formation rests conformably upon the Lamotte sandstone without any sharp line of demarcation, the lower part of the Bonneterre consisting of a transition zone composed of alternating, thin-bedded and sometimes cross-bedded

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[3] - Winslow, Arthur Vol. VI Chapter VII. of the earlier reports of the Missouri Bureau of Mines and Geology.

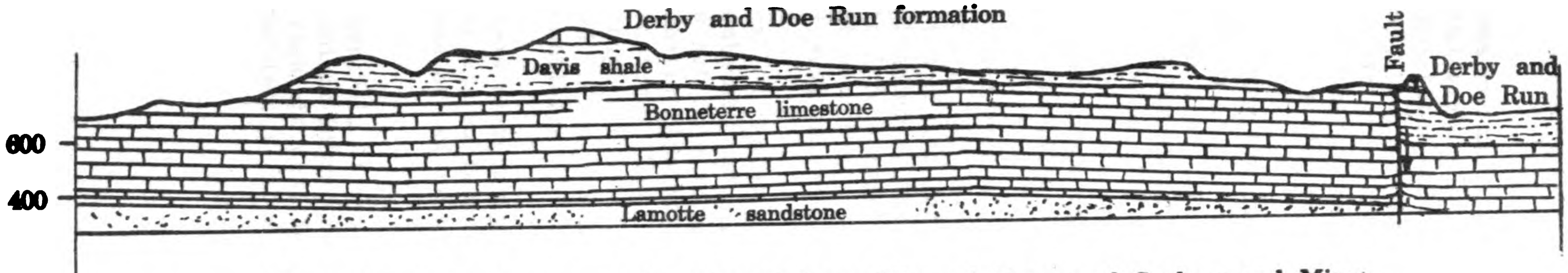


Figure 1. – General cross section adapted from, Missouri Bureau of Geology and Mines, Vol. IX, part 2

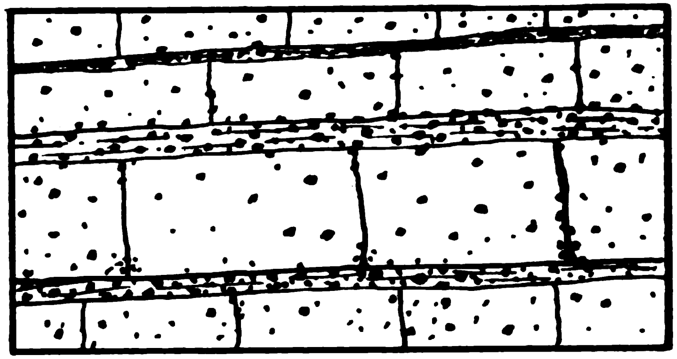
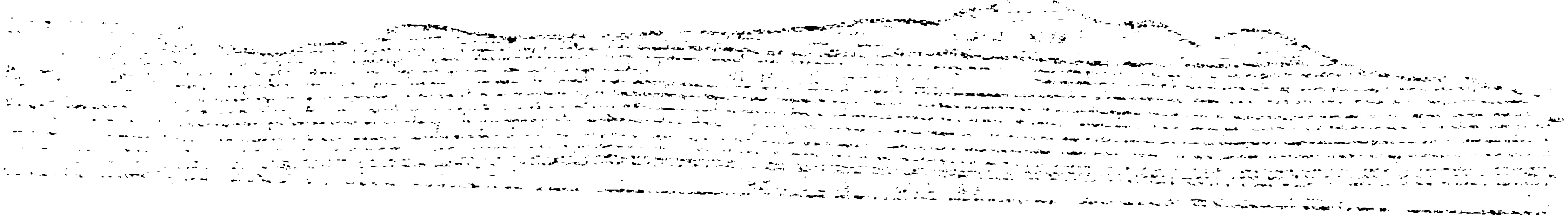


Figure 2. – Typical mineralization in a 10-foot face of ore



layers of limestone and sandstone interbedded with thin layers of "chloritic" or glauconitic material. The Lamotte sandstone locally carries disseminated lead near the top and sometimes extending to considerable depths below the Bonneterre contact. No ore is being produced from the Lamotte, however. Figure 1 is a generalized section in the vicinity of the mine which is the subject of this paper. The top of the Lamotte is very uneven, causing abrupt changes in the level of the Bonneterre floor. The formation is very porous and is saturated with water.

Considering the district as a whole, it may be said that the ore is segregated principally in the lower 100 feet of the Bonneterre, although it occurs at various horizons from the top to the bottom. The galena appears to favor the darker, coarse-textured limestone beds and the more shaly parts of the formation. The shale layers, which are usually thin, are often highly impregnated with galena while the lime beds on either side are only lightly sprinkled with the mineral (Fig. 2.). As a rule the richest rock is soft and somewhat decomposed, though there are a few notable exceptions to this.

The upper part of the Bonneterre is often somewhat argillaceous. Overlying this formation is the Davis shale, which has a normal thickness of 150 to 190 feet. Between the top of the Bonneterre and the bottom complex of limestone, shale, sandstone, and glauconite, the formation is thicker bedded, the bedding planes being commonly 1 to 6 feet apart. Vertical jointing in this horizon is often strong, but the joint planes are usually 3 to 5 feet or more apart. Joint planes are sometimes filled with galena and calcite.

So called "water channels" cut across the beds in various directions but strike predominantly in a general easterly and westerly direction. These channels some times start from surface or in the overlying shale and pass into the limestone, and may gradually play out at depth. Others pass from the Bonneterre into the underlying sandstone. The larger water channels seldom carry ore in commercial quantities, although sheets of galena and aggregates of crystals are often found in them embedded in reddish or brownish mud and decomposed dolomite, the usual filling of these channels.

Systems of roughly parallel fracture zones occur in the upper horizons, individual fractures being thin, leached, and whitened by the action of surface waters. These are often mineralized and frequently have workable ore associated with them.

In general, it may be said of the district that galena occurs disseminated through dolomitic limestone and shaly beds, in horizontal sheets along bedding planes, in vugs and cavities, filling or lining the walls of joints and crevices, and as aggregates of cubes in channels and joints.

The greater part of the production from this particular property is now coming from the middle and upper horizons of the Bonneterre formation, although the ore occurs over a vertical range extending from within a few feet of the surface to a depth of 450 feet. Mining operations are being carried on at various horizons from +376 to +685, sea level datum, and ore has been mined to elevation + 771.

The thickness of the individual ore horizons ranges from 7 feet (the minimum mining height) to over 200 feet. At present, however, there are no active stopes over about 35 to 40 feet high. In a given area the ore may occur at several overlapping horizons or may be confined to only one or two. (Fig. 3 and fig. 4.)

CHARACTER OF ORE AND WALL ROCKS

In the middle horizons the galena usually occurs in rich horizontal layers a few inches to a few feet in thickness, while the intervening beds are sprinkled lightly with the mineral. Thus the ore body is composed of layers of rich ore separated by lean ore rather than of bands of ore separated by barren rock. In an ore area all the rock contains some galena. The ore of the middle horizon is harder and breaks in larger blocks than that in the lower and upper horizons due to the heavy bedding and jointing, the beds and joints being closer together in the latter. The ore in the southern part of the district occurs largely in the lower part of the formation and on the whole breaks finer.

In the upper levels of this mine are numerous systems of slips and crevices, usually leached white, from which layers of workable ore make off along the bedding and fracture planes. Some of these layers are quite rich.

The outlines of the ore bodies are very irregular both in plan and section, as shown in Figure 2 and Figure 3. Stoping widths range from a few feet along some of the upper narrow fractures to several hundred feet. Some of the ore bodies are 700 to 800 feet wide by over 1200 feet in length.

There is considerable variation in the strength of the formation in different localities, but on the whole it is very strong and will support itself over wide spans. Some spans are over 100 feet between pillars or margin walls. In other places it is necessary to limit spans to 17 or 18 feet. Shaly layers tend to slack upon exposure, and vertical channels, slips, and joints constitute other sources of weakness in both ore and roofs of stopes. "Water channels" constitute a condition that requires care in mining, both because they weaken arches and spans and because they are sometimes active watercourses. In some parts of the district care is taken to leave barrier pillars against these channels to prevent a heavy influx of water from them.

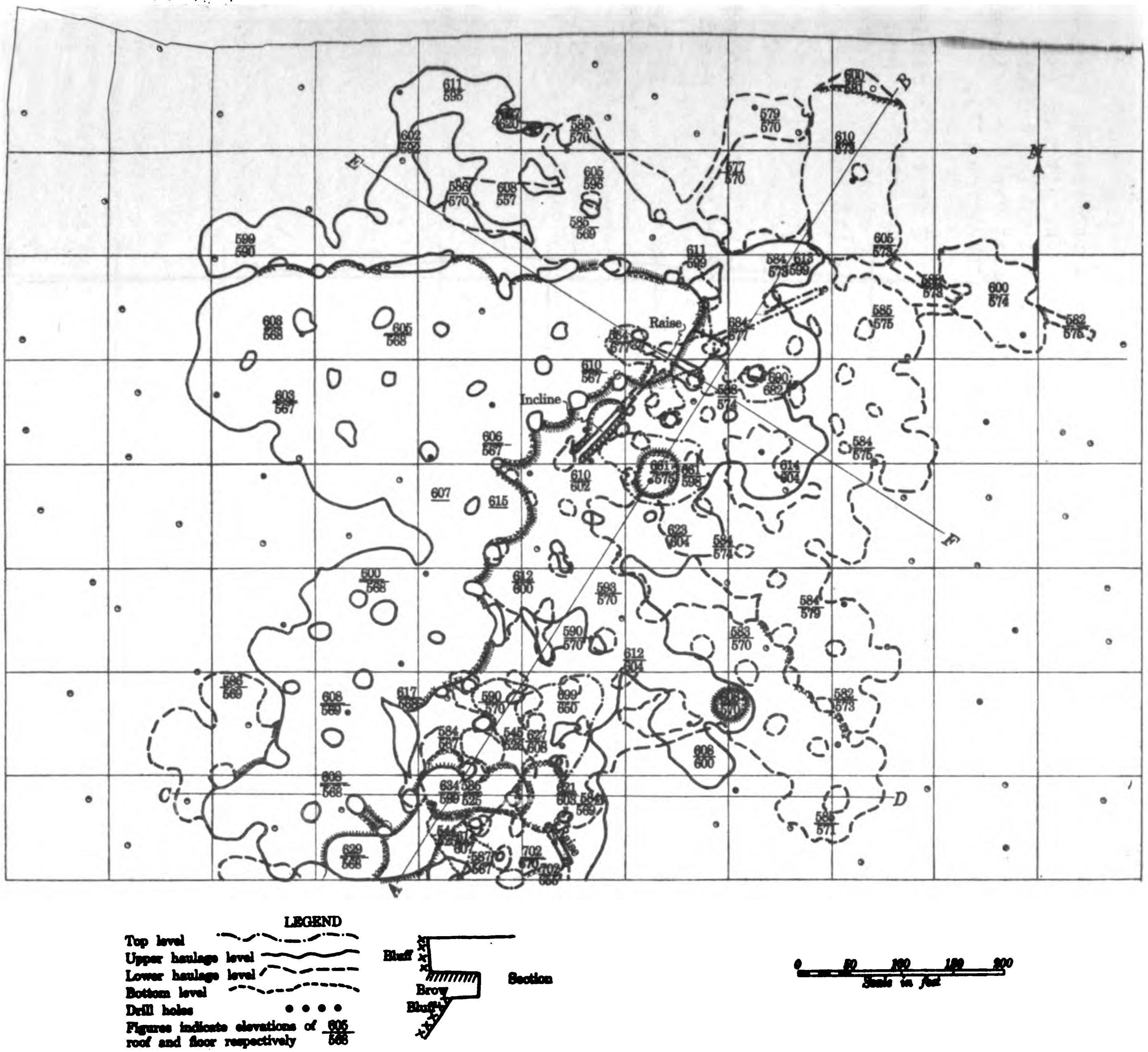


Figure 2.– Plan of portion of mine workings

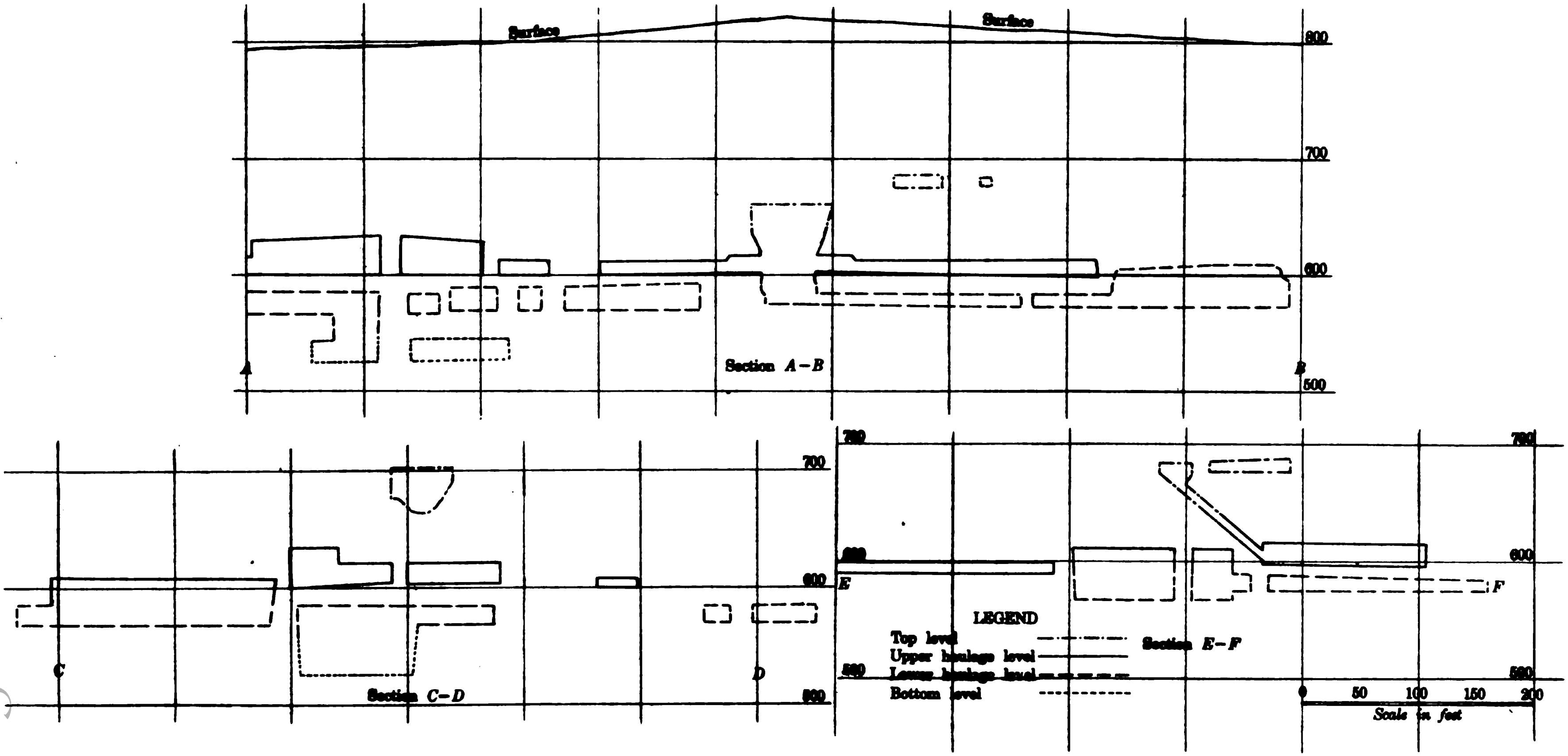


Figure 4.—Sections through mine

The wall rocks are in general of the same character as the ore; and the ore horizons pass laterally, sometimes gradually and sometimes abruptly, into barren rock.

PROSPECTING AND EXPLORATION

Exploration is conducted almost entirely by diamond drilling from the surface, diamond drilling underground, and test drilling with light hammer drills, rarely by means of prospect drifts.

Diamond Drilling. The company owns and operates its own diamond-drilling equipment using on surface "B" size rods, core barrels, and bits making $1\frac{1}{4}$"inch cores and "E" rods with "A" bits and core barrels underground making 1-1/8-inch cores. All cores are carefully examined by an inspector who, through many years of experience in the district, can estimate very accurately the lead content of the rock. Notes are made regarding the physical character of the core, its color, porosity, the estimated lead content, etc., and are carefully recorded. The cores are marked and filed for future reference. These cores are not assayed. The sludge from the drill holes is dried and assayed, and the assay results are recorded with the other information.

The diamond drill is used on the surface chiefly for prospecting areas outside the boundaries of the old stopes and underground to test floors of old stopes for ore which may have been left in early years and for tests holes too long to drill with the jackhammer, which is limited here to depths of 22 feet. One diamond drill is kept in operation on both day and night shifts all the time and is moved from one section of the mine to another as required. The underground drill will average 40 to 50 feet of hole per shift.

Test-Hole Drilling. Light jackhammer-type drills are employed for testing backs of old stopes, floors, and marginal areas where the depth of hole required does not exceed 22 feet. In active stopes they are operated from the bottom of headings to test the back before stoping up the bluff or bench. For testing old backs in high stopes they are operated from scaffolds suspended from the roof. These scaffolds are a unique feature of the practice in the district, and as they are employed for other purposes they are described later under the general caption "Mining". For drilling other than down holes the drills are mounted on a column and are equipped with a pneumatic feed, which is another unique feature of the practice at this property; this is described later under "Drilling and Blasting". When operated from a scaffold for testing the roof they are sometimes mounted on a bar secured to the hanging rods, and the drill is counterweighted by passing a rope tied to the drill over a pulley in the roof and attaching one or two old car wheels to the other end of the rope. It is customary to test backs with holes drilled at the corners of 25-foot squares over the area to be explored.

The drills usually are operated dry and the cuttings are caught in a pan which fits around the front end of the drill. The steel is made up in 2-foot chang starting with a six-pointed rose bit, which has a gauge of 1-3/4 inches. The gaug diminishes 1/16 inch with each change of steel up to the 10-foot length, after whic chisel bits are employed with gauges calipered to suit the hole. The cuttings fr each 2 feet constitute a sample, which is boxed and tagged; the samples are turned in to the office upon going off shift. These samples are assayed for lead and the values recorded.

A large part of the present output of this mine is produced in areas which had been worked over in early years and at that time presumed to be mined out. Subsequent exploration, as described above, has resulted in the development of much or over the backs, in the floors, and outside the margins of the old stopes.

In determining the minability of an area explored by these methods the physical characteristics of the core, such as color and texture, are considered in conjunction with past experience as to values at the same vertical horizon of the formation in adjacent areas, as well as the lead content indicated by the drill samples. Thus a single hole showing good values at a given elevation, although surrounded by holes showing low values or only traces of lead at this elevation, is usually justification for the development of an area provided adjacent areas at the same horizon have been productive.

ESTIMATION OF TONNAGES AND VALUES

In estimating ore proved as a result of drill exploration a minimum mining thickness of 7 feet and 15 feet-per cent lead are taken as a basis for the calculations. That is to say, ground which will not average 15 feet-per cent over a 7-foot thickness is excluded from the ores reserves. Under this rule 1 foot of 15 per cent lead would be included, and as a matter of fact would mine out better than the minimum grade, since the 1 foot of high grade has a much higher specific gravity than the other 6 feet of barren ground which must be included under the rule requiring a minimum stope height of 7 feet. Furthermore, experience has demonstrated that all the rock within the boundaries of an ore body contans some lead.

Where more than 7 feet are involved a calculation is first made to determine whether the entire thickness is minable, that is, of an average grade of 2 per cent lead. If it is not, the minable thickness and grade thereof are determined by a cut-and-try method.

DEVELOPMENT

This property extends over a large area and embraces several of the oldest mines in the district which were originally developed as separate operations.

There are 21 shafts on the property that were sunk to develop a number of separate ore bodies, which were in some cases widely separated by barren ground. The several productive areas have since been connected underground by crosscuts, some of which are long, and now all the ore mined is hoisted through a single shaft, designated as No. 1 shaft. Ore is hoisted from two levels located, respectively, 175 feet and 245 feet below the collar. It is located at the mill site.

This shaft is 6 by 15 feet in the clear and is divided into three compartments. Two compartments are used for hoisting in 2 3/4-ton skips in balance, and the other is used as a pipeway and for power cables. There is no ladderway in this shaft.

Close to No. 1 shaft and near the change house and office is No. 1 man shaft, a shallow shaft provided with a stairway from the collar to the bottom. From the bottom of this shaft access to the upper main haulage level is gained by stairways leading down through old stopes.

No. 3 shaft is equipped with a single cage and counterweight and is used for handling men, explosives, and supplies. It is connected to both of the main haulage levels.

Another shaft equipped with a cage and hoist is used as a ready means of access to a large pump station located in an isolated and inactive section of the mine. There is no regular attendant at this station, the pumps being automatically started and stopped. Yet another shaft is used for lowering heavy equipment and material which can not be conveniently handled in No. 3 shaft. The bottom part of this shaft is untimbered and is, in fact, just a high open stope. Other shafts merely assist in providing excellent natural ventilation, while still others are abandoned and closed.

Practically no timber is required to support the ground in any of these shafts, except right at the surface, and the timbering used is only that required for support of guides, pipes, etc.

As stated, much of the present production is from operations in areas previously worked, where subsequent exploration has disclosed ore bodies over, beneath, and outside the old stopes. Ore is now being mined from elevation +376 to +636 on some 10 different levels. There are two main haulage levels 70 feet apart vertically, the upper one being at elevation +625 and the lower at + 555 sea level datum, at the hoisting shaft. About 75 per cent of the present output is either dropped down to these levels through chutes or is hoisted on slopes. To connect with the upper ore bodies it is often necessary to drive crosscuts in waste to get under them and then raise thereto.

In some cases access to these stopes is from upper levels or old stopes, and in others it is necessary to divide the raise into two compartments, one for ore and the other for entrance to the stope and for air and water lines. Sometimes where an ore body is small and the cost of development per ton of ore to be recovered would be excessive it is more economical to tram the ore to an old stope connecting with a haulage level and reload the ore with a mechanical shovel after enough tonnage has been dumped to warrant moving in a shovel.

All waste from development is trammed and dumped into old stopes, no waste rock being hoisted to surface. It is obvious that under the conditions at this mine, where most of the development is in old areas, no standard method of development can be employed; rather the problem is to fit the development in each particular case to the existing workings.

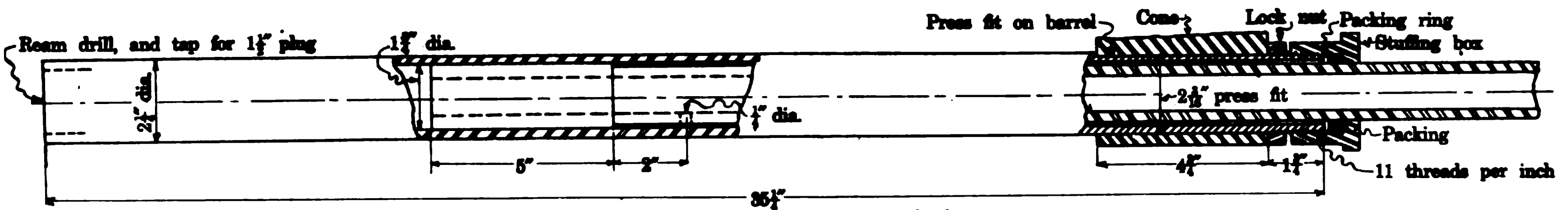
Development openings and haulage drifts are untimbered. Main haulage drifts are a minimum of 8 by 12 feet in cross section with the track usually in the center, a ditch on one side, and room to walk on the other.

Development drifts and crosscuts are usually $7\frac{1}{2}$ or 8 feet high by 9 to 10 feet wide. Raises are about 6 by 9 feet and are commonly driven at an inclination of 45 degrees, seldom vertically.

Slopes are driven to develop ore horizons below the haulage levels, and the inclination varies up to a maximum of about 30 degrees from the horizontal. In cross section they are about 6 to 7 feet by 10 to 12 feet. From some of these slopes several levels are turned off to develop as many different ore horizons.

DRILLING AND BLASTING

All drilling is done with light jackhammer drills operated dry, except in confined places where the dust is not readily dissipated. As there is practically no silica in the rock the dust is not considered injurious. Furthermore, the drills are commonly mounted on a pneumatic feed described later, so that the miner stands back from the drill while it is in operation. The drills are all operated as one-man machines, except in scaffold construction work. Air pressure at the drills is maintained at 75 to 80 pounds.



Detail of barrel and assembly of pneumatic feed, barrel of steel tubing

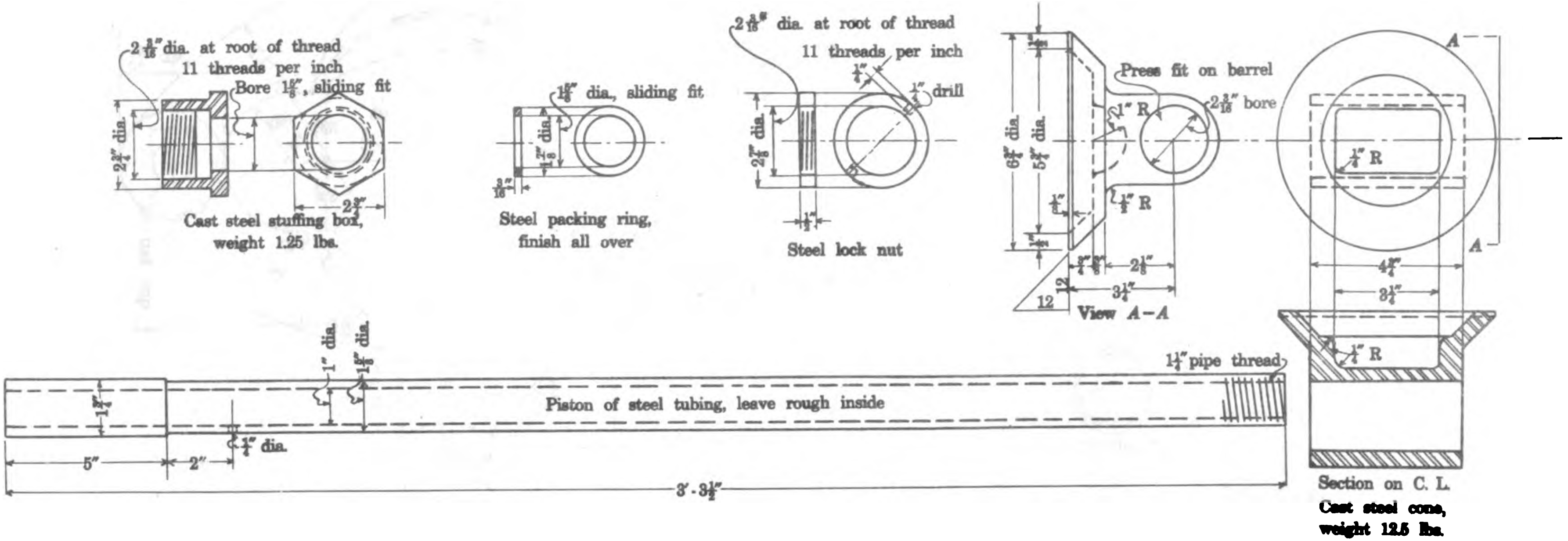


Figure 6. – 1.75 inch pneumatic feed for pneumatic hand drill

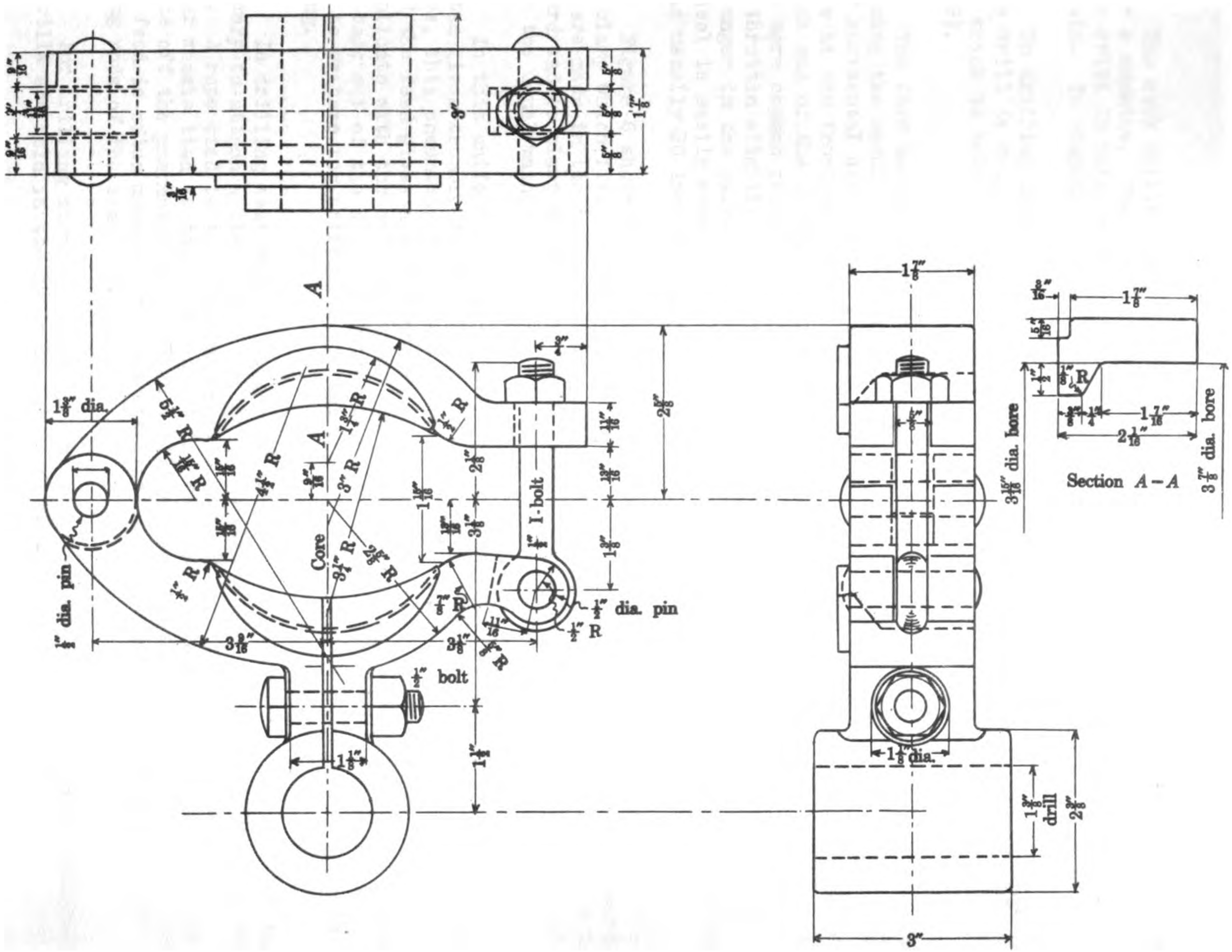


Figure 7.– Clamp for drill. Cast steel, weight 12 pounds

The rock drills easily, and 80 to 100 feet of hole per shift is a fair average for a machine. The steel holds its gauge well, and it is usual in development work to drill 12 holes with one set of steel; even then much of the steel could be run again. In stoping the steel is run until the gauges will not follow.

In drilling drifts, raises, stope leadings, and prospect holes in roof or rib the drill is mounted on a pneumatic feed plunger operating in a cylinder or barrel which is held in a saddle clamp attached to a 3 inch, single-screw column (fig. 5).

The feed barrel can be rotated through 360 degrees in a vertical plane, and by turning the saddle clamp about the vertical column bar the machine can be pointed in any horizontal direction. The air connection for the pneumatic feed can either be made at the front end of the piston or plunger close to the drill machine or at the back end of the barrel, according to the preference of the operator. The former is the more common practice. By shutting off the air to the feed and opening the drill throttle slightly the machine backs the drill out of the hole. By rotating the plunger in the barrel, thus swinging the machine out of line with the hole, the steel is easily removed and the next steel inserted in the hole. The length of feed is usually 30 inches, though some have been constructed with 33-inch feed.

Figure 6 shows the construction of the barrel, piston, and cone and Figure 7 the clamp which fits around the drill shell and secures it to the piston or barrel. These are made in the company shops. The saddle clamp in which the cone fits is the standard saddle clamp made by the drill manufacturers for mounting the drill on an arm in the usual manner.

In this uniform cutting ground, which is comparatively free from slips such as often given trouble in drilling and where there is little tendency for the holes to cave, this pneumatic feed gives exceptionally good results. The air pressure behind the feed gives a steady pressure against the rock which would be difficult to duplicate with the ordinary screw feed. After starting the drill the miner can stand back out of the dust or sludge and can occupy himself with claning off the bench to prepare for drilling stope holes or with other work while the machine is cutting.

In drilling test holes in the roof the same equipment is used. Here it is customary to attach a light clamp to the top of the column to which a pulley is secured. A rope tied to the drill and passed through the pulley, with one or two old car wheels tied to the other end of the rope, serves to take the weight of the machine off the pneumatic feed. When drilling these holes from a scaffold the pneumatic feed is often mounted upon a short cross bar fastened between a pair of the hanging rods of the scaffold.

For drilling stope holes and other holes approximately vertically downward, the drills are held in the hands in the usual manner.

For drilling "lifters" in the stopes or other flat holes along the floor it is customary to use a "drilling board" upon which the drill rests. This consists of two parallel strips of wood with slats nailed across the underside to form a ladder. These strips are about 2 inches wide, with spaces of 2 inches between them. The "board" is laid flat on the floor in the direction of the hole to be drilled, and the drill is rested upon it. The space between the side boards is such that the drill slides smoothly upon them. After starting the hole the driller exerts a forward pressure against the back end of the machine by means of a pinch bar inserted between the slats and with a slat as a fulcrum. He is thus able to stand erect while feeding the drill forward.

The machines employ 7/8-inch, hollow, collar-shanked steel made up in 2-foot changes up to 22 feet in length. The six-point rose bit is used with a 1 3/4-inch starter and gauge changes of 1/16 inch for each change of steel. The steel is heated in a coal-fired forge and is machine sharpened in a shop on surface at No. 3 shaft. The drillers handle their own steel from shaft to working face.

Each driller is provided with a tool box in which he keeps his tools, a few spare fittings, gaskets, etc., and a detonator box.

Electric detonators in delays 1 to 6 are employed for all development work. For stoping and block-holing fuse and caps are used. Where electric detonators are used the miner is supplied with a box containing a powder punch, a bundle of tarred rough twine, a roll of connecting wire, and a galvanometer. Where fuse is used a similar box contains a powder punch, a bundle of tarred rough twine, a cap crimper, a fuse cutter, a can of waterproofing compound, and a reel mounted inside the box at one end, on which a roll of fuse is wound. The loose end of the fuse is drawn through a hole on the reel end of the box. Over this hole and on the outside of the box a bracket is fastened which is a part of the fuse cutter and holds the cutting element. The latter is kept in the box when not in use.

Each miner prepares his own fuse and primers, locally termed "nifters", and loads and fires his own holes. A special powder punch is used as illustrated in the sketch, Figure 8. This consists of a brass tube closed at one end into which a stick of powder fits easily but snugly and down the center of which is the punch proper. The cartridge is pushed back to the end of the tube and is then withdrawn with a hole punched exactly in the center and half the length of the cartridge. The capped end of the fuse is inserted in this hole and pushed to the bottom. A piece of twine is then looped about the fuse close to the end of the cartridge and a hitch around the center of the cartridge holds the detonator in place, so that it will not pull out. The nifter is placed in the hole with the fuse end pointing out so that the

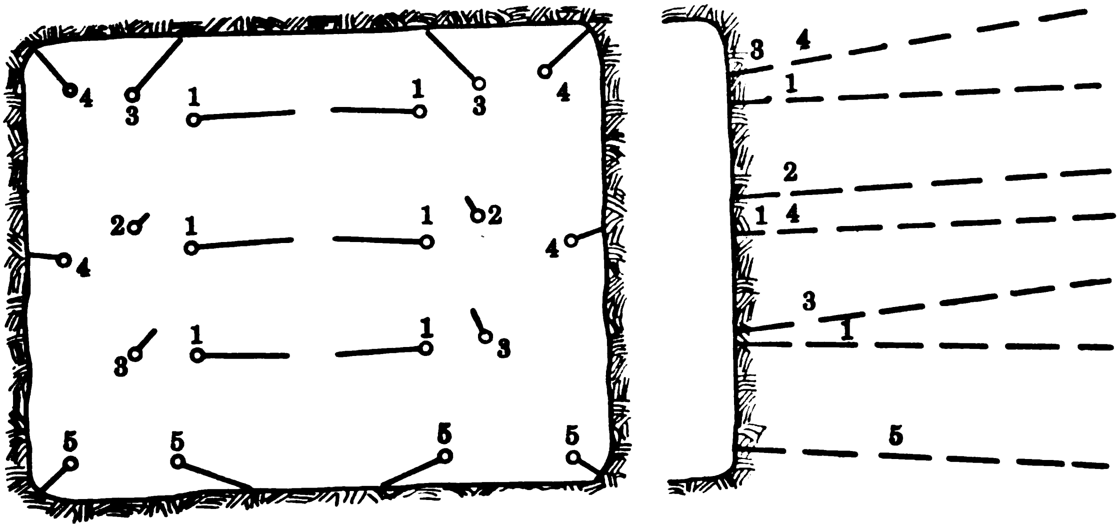


Figure 9.—20-hole center-cut round in 8′×10′ drift

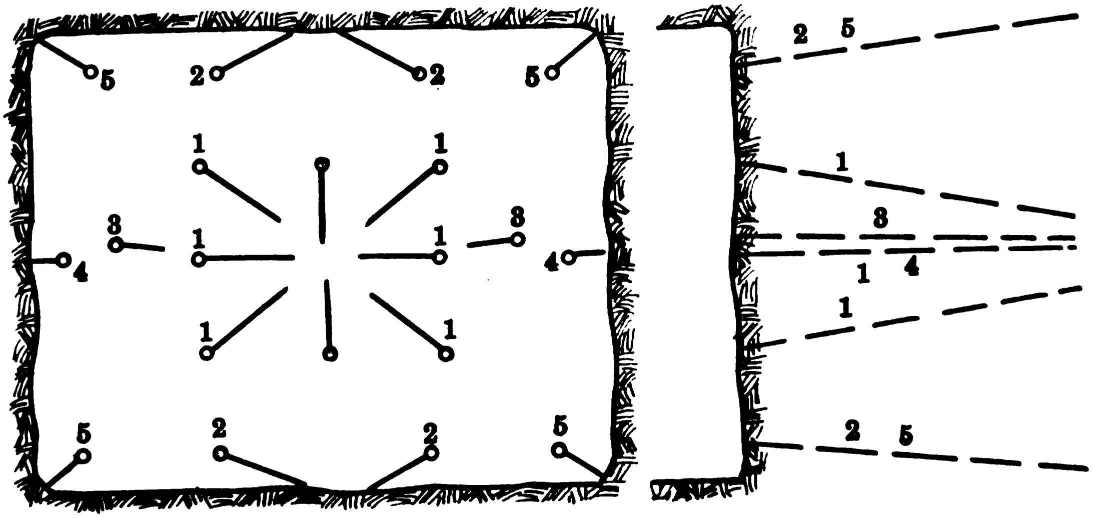


Figure 10.—Pyramid or dishpan cut round in 8′×10′ drift

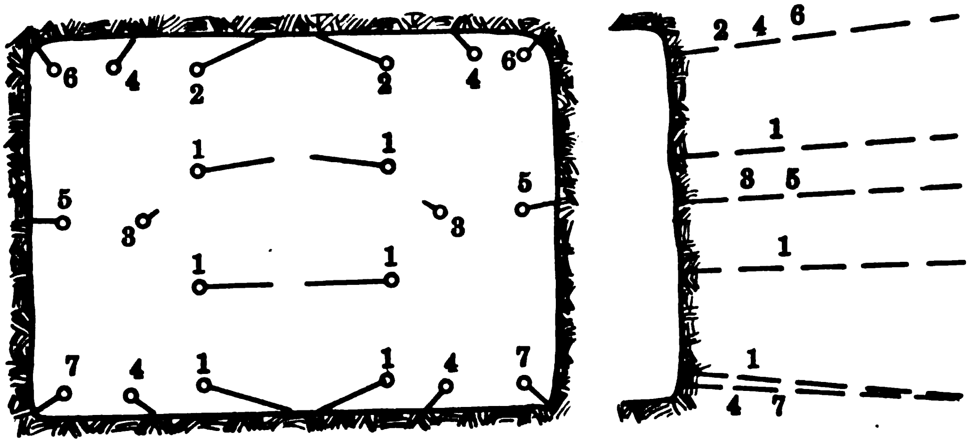


Figure 11.—20-hole center-cut round in 6′×9′ raise.
Numbers show order of firing

use is not bent. The rule is to place the "nifter" next to the last stick in the hole, although this practice is varied and it is sometimes placed near the bottom.

The explosive is a 40 per cent ammonia gelatin running about 100 sticks to the 50-pound box. Stemming is prepared in cartridges by the miner from drill cuttings and two or three stemming cartridges are placed on top of each charge.

Where electric blasting is employed current is usually taken from the 220-volt mine circuit. In rare instances, blasting machines are used in remote sections of the mine where power is not accessible.

Two switches are employed, an ordinary knife switch which is mounted within sight of the face to be blasted but far enough back to escape injury from the blast and a firing switch mounted some distance away in a locked box. The firing switch closes against a strong spring, which keeps it open except when held in the closed position.

Before connecting the lead wires the miner is required to insert an electric squib across the ends to test them for stray currents. If the squib does not fire he closes the knife switch and throws in the firing switch. The burning of the squib indicates that the circuit is all right up to that point. The trip back to the firing switch assures him that it is disconnected and in working order. Going back to the face, he throws out the knife switch, removes the burned squib, which he retains, and then connects the lead wires to the connecting wires. The knife switch is thrown in and then lastly the firing switch. The burned squib is attached to a card provided for the purpose, upon which the driller marks the date, place, and his signature and turns it in to the office at the end of the shift as evidence that he has complied with the regulations of the company as outlined above.

In driving drifts, crosscuts, raises and slopes, the round usually employed is a "V" or center-cut round. Figure 9 shows a round employed in an 8 by 10 foot drift and Figure 11 a round in a 6 by 9 foot raise driven at an inclination of 45 degrees. The round is varied somewhat to suit the ground and according to the judgment of the miner. Most of the drillers are men of long experience and unusual intelligence and can usually give very good reasons for the particular variation of the round which they are using in any special instance.

The drift round shown was in somewhat easier breaking ground than the average, pulling 6 to 7 feet, and was broken with about 125 sticks of 40 per cent special gelatin. The cut holes are loaded with about 8 sticks each and the relief and square-ups with 4 to 6 sticks each. A perhaps more common round employs 8 cut holes, the bottom lifters of the illustration being drilled as cut holes and with additional relief holes as required. There are usually 22 holes in the average round, and 24 to 26 holes are not uncommon.

The 20-hole raise round shown in Figure 11 is similar to the drift round. Six to eight sticks of powder are used in each of the cut holes and an average of 4 sticks in the other holes.

Where there is pronounced horizontal bedding, especially if there are alternating layers of lime and shale, a modified pyramid cut (locally termed a "dishpan" cut) has been found to give better results than the "V" cut. This cut is shown in Figure 10, the position and number of relief and square-up holes depending upon size of drift and ground conditions.

The average advance in drifts and raises is about 28 feet per six-day week. The miners work in pairs with two, column-mounted, pneumatic-feed, jackhammer-type drills on one shift, the round being mucked out on the opposite shift. Development headings are driven on contract at a given price per foot of advance.

STOPING

Open stopes with pillar support are employed for extraction of the ore, the pillars being of the casual or irregular type as contrasted with those employed in regular room (or stope) and pillar work.

Where the ore is not more than 9 or 10 feet thick it is breasted out the full height. In thicker ore a heading 7 to 8 feet high is breasted out immediately under the top of the ore and in advance of the stope which follows it up closely. The method of breasting out thin ore is practically the same as that of driving headings in higher ore, as illustrated in Figure 12, which also shows the method of drilling the stope or "bluff" ore.

In the heading three holes are drilled one above the other, but the middle or breast hole is drilled with about 6 inches less burden than the back hole and lifter. The burden is usually 3 to 4 feet, although this varies somewhat with the nature of the ground. These holes are drilled 8 to 10 feet in depth, and occasionally deeper. In the heading, the breast hole, usually loaded with about 10 sticks of powder, is timed to go first, followed in turn by the back hole and lifter, which are loaded somewhat lighter. The face of the heading is not carried in a straight line but as shown in the illustration in order to provide free faces to break to.

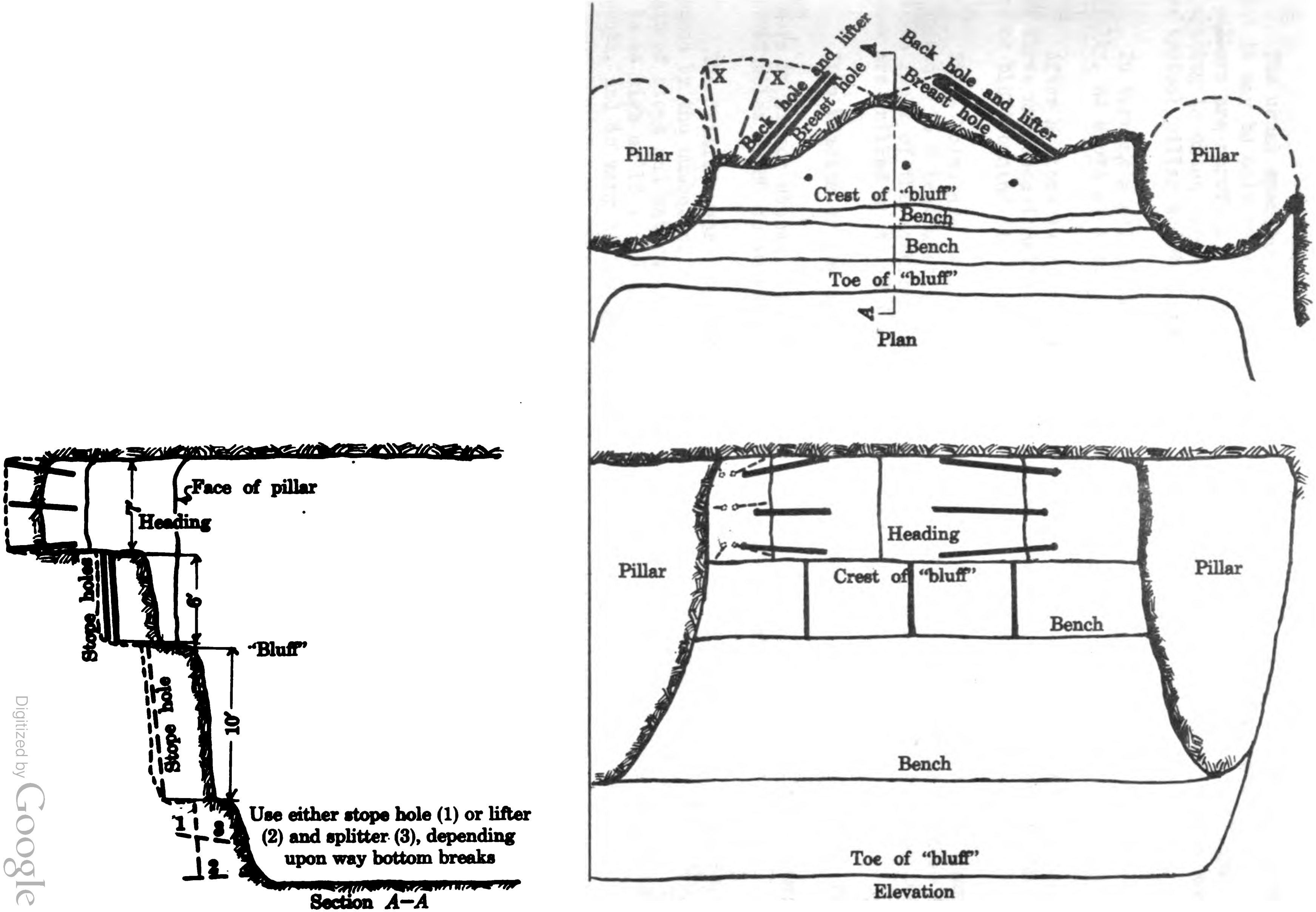


Figure 12. – Heading and stope or "bluff"

The usual span between pillars in good ground is 30 feet, but in weaker
und it may be only 16 to 17 feet. Where there are old workings above or below
pillars are carefully located by survey to "match" with the older pillars; thus
spacing is often very irregular. The plan, Figure 3, is drawn to scale and
ws typical pillar spacing and arrangement.

In turning a pillar the holes are never drilled toward the pillar but tan-
ıtially, as shown at x-x, to avoid possible shattering of it.

After a stope is opened there are usually two headings in operation, some-
ıes three and occasionally only one. One driller works in each heading and the
pe or bluff behind it.

Stope holes are drilled 3 to 4 feet back of the face of the "bluff" and
spaced 6 to 8 feet apart, depending upon the nature of the ground. Those drilled
m the floor of the heading are 6 feet deep, while those drilled from the lower
nches are drilled up to 10 feet in depth.

In a heading or low stope a driller will usually break about 30 tons per shif
d more in higher stopes. In "bluff" mining 90 to 120 tons per shift is common.
out 12 holes or 80 to 100 feet of drilling is an average shift's work.

The drillers work on contract and are paid for the tons of ore broken, as
asured by the number of cars of 2½ tons capacity. The price varies with the
ight of stope and nature of the ground. In low stopes and hard ground the price
y be as high as 16 to 18 cents per ton, for stopes of intermediate height 12 to
cents, and in very high stopes as low as 8 cents.

In the low stopes and headings the ground usually breaks fairly small, but
the high stopes or bluffs considerable secondary breaking is often required.
wlders requiring block-holing can usually be pushed to one side by the mechanical
ovel to be drilled and blasted at the end of the shift.

LOADING

In development work and in stopes where enough ore can not be broken to
eep a mechanical loader busy the ore is shoveled into cars by hand. The usual
ask or "score" for a hand loader is 20 tons. This score is reduced, however, where
oading is difficult, as when there is a long tram. The mucker does his own tramm-
ng from the siding or "loop", as it is locally called, to and from the loading
oint, ordinarily using a mule.

At present there are seven mechanical loaders in the mine. These are of t shovel or "dipper" type, mounted on caterpillar traction, and have a full 360-degr swing; the dipper discharges its load directly into mine cars. These shovels are two different types. One is equipped with a single 25-hp., A. C. motor, all three motions being operated through clutches to the same drive. The other has separate drives for each motion. This shovel is described in detail by Van Barneveld.[4]

The shovels are operated by one man, who usually also transfers his own ca from the locomotive gathering loop to the shovel, using a mule. The shovel operat is paid for loading under a bonus system whereby a "score" or task is set for whic he receives a regular daily wage, and additional payment is made for tonnage in excess of this. The usual score is 42 cars of 2½ tons capacity each, but this is var to suit special conditions. Occasionally in extreme cases it is as low as 32 cars, or as high as 48 cars.

Two muckers are usually employed in each development heading and are given 8, 10, or 12 hours pay for mucking out the round. This amounts to a bonus for leaving a clean set-up for the drillers working on the opposite shift.

STOPE SUPPORT

The stopes are supported on irregular circular pillars, and no timber is us for this purpose. Pillars are commonly spaced with an intervening span of 30 feet, sometimes as little as 17 or 18 feet, and in some of the older workings the spacing is 40 - 50 and even 80 feet or more. (Figure 3.) It is estimated that 10 per cent of the stope areas is left in pillars.

In high stopes the old pillars are often very irregular in shape and size from top to bottom, due to differential weathering of the lime and shaly layers or to slabbing along slip and joint planes. The shaly layers tend to slack and crumble, often leaving pillars shaped like an hourglass or several superimposed hourglasses. Vertical jointing in the rock also serves to weaken pillars, and where a prominent joint is close to a pillar face a large slab may scale off.

These conditions have made it necessary in many instances to reinforce pillars when their continued maintenance is required. In some instances gunite has been applied to stop slacking action. In others it has been necessary to enlarge the pillars by building forms around them and filling with concrete, and in still others pillars are wrapped with old cable over vertical rods, like the hoops n a barrel. Another practice is to drill horizontal holes through the pillar through which heavy tie rods are employed to hold the pillar together.

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[4] - Van Barneveld, Charles E., Mechanical Underground Loading in Metal Mines; U. S. Bureau of Mines, and University of Missouri School of Mines and Metallurgy. 1924. pp. 142-152.

Some of the stopes are very old, and in one section of the mine pillars over 00 feet in height are standing. The active stopes are low in comparison, very few eing over 35 to 40 feet high; but inasmuch as they are often above or below old topes and the pillars are "matched" with older pillars, they constitute in effect, ong continuous pillars with the floors acting as beams between them.

SCAFFOLDING

A unique feature of the practice in the Southeastern Missouri lead belt is the use of scaffolding suspended by hanging rods from the roof of the stope for nspection and trimming of roof and pillars, for drilling test holes for exploration purposes, or for gaining access quickly and cheaply to ore otherwise accessible with difficulty in order to open it up for development and mining.

In high ground the active life of many stopes has been very long, and although the ground is very firm and strong the roof may become weakened along bedding planes, shaly layers, slips, and channels due to long exposure to the air. The scaffold affords a quick, cheap and safe method of access to the roof for inspection and for trimming down loose ground.

In strong rock similar to this the tendency when failure does occur is for the ground to come in big blocks. The use of scaffolds as constructed and employed to-day is the outgrowth of an incident some 12 or 13 years ago. A large slab of loose ground appeared on the side of and at the top of a pillar about 100 feet high in an active stope. In the roof above it another loose slab opened up so that work in the stope had to stop. The pillar stood about 40 feet from the rib of the stope, along which was a traveling road. A 3-inch plank 18 feet long was run out from the traveling road and jacked against the back with two drill columns. A miner then went out part way on the plank and drilled holes in the roof, in which eye bolts were inserted. From these hanging rods were dropped to support the plank. Holes were drilled further ahead and the scaffold extended till the loose ground was within reach, when it was safely barred down. The present practice has become pretty well standardized, and the men have become expert and rapid in this work. The men work in pairs and secure themselves by ropes fastened around the body and fastened at the other end to a solid anchorage. Two men will often extend a scaffold 50 to 70 feet in a shift; in fact, 100 feet has been done.

In starting a scaffold access is gained to the top of the stope by means of ladders or a roadway along the side of the stope. An anchorage is made in the rib or floor of the roadway, and two holes are drilled ahead in the roof of the stope about 3½ feet apart, the line between them being at right angles to the intended direction of the scaffold. These holes look ahead at an angle of about 45 degrees and are drilled to a depth of 14 inches. (Figure 13.) Eyebolts of 1-inch round wrought iron with ends split to receive a wedge are inserted in the holes and driven up tightly. Hanging rods of 1-inch round iron 7 feet 3 inches in length are then hooked into the eyebolts, and a cross bar, also of 1-inch round iron, is hooked into the eyes at the bottom ends of the hangers.

Two 2-inch planks 8 feet in length are secured to the crossbar by U bolts of ½-inch iron which pass through holes bored about 6 inches from the ends of the planks. The planks are then pushed ahead to the position shown, the hangers then being at an angle of about 45 degrees from the vertical. The planks are anchored at the rear end, as shown in the sketch, or may be secured to horizontal anchor rods by means of U bolts.

One driller then stands on the outer end of the scaffold and guides the drill in starting the next pair of holes, while the other operates the drill from the floor of the scaffold further back. The operation is repeated, the rear ends of each pair of planks being secured by U bolts which are long enough to pass through two planks, one on top of the other, as shown. The roof is of course tested and loose ground barred down as the scaffold advances.

Some of these scaffolds are several hundred feet long. Changes in direction can be made as desired, and often they are jumped from one elevation to another either up or down to suit changes in the elevation of the roof.

The scaffolds are often guyed or braced to stope walls, roof, or pillars to prevent side sway.

TRANSPORTATION

Main haulageways are laid with 56-pound rails to 24-inch gauge on closely spaced well-tamped wooden ties. The roadbeds are well ballasted, granulated slag being used for this purpose in some sections. The curves are easy, the rails are braced with standard steel rail braces, and the outer rail is properly elevated. Long switch points are employed and are connected to low, cross-throw switch stands.

On haulageways over which more than one locomotive travels semiautomatic block signals are installed, which are operated by the locomotive engineer from his seat while the train is in motion. At some other mines of the company full automatic block-signal systems have been installed. All switches are protected with automatic lights which show red for the turnout or for a split switch. The lights are in pairs, so if one burns out the other will show. This equipment is carefully inspected and maintained in perfect working order at all times.

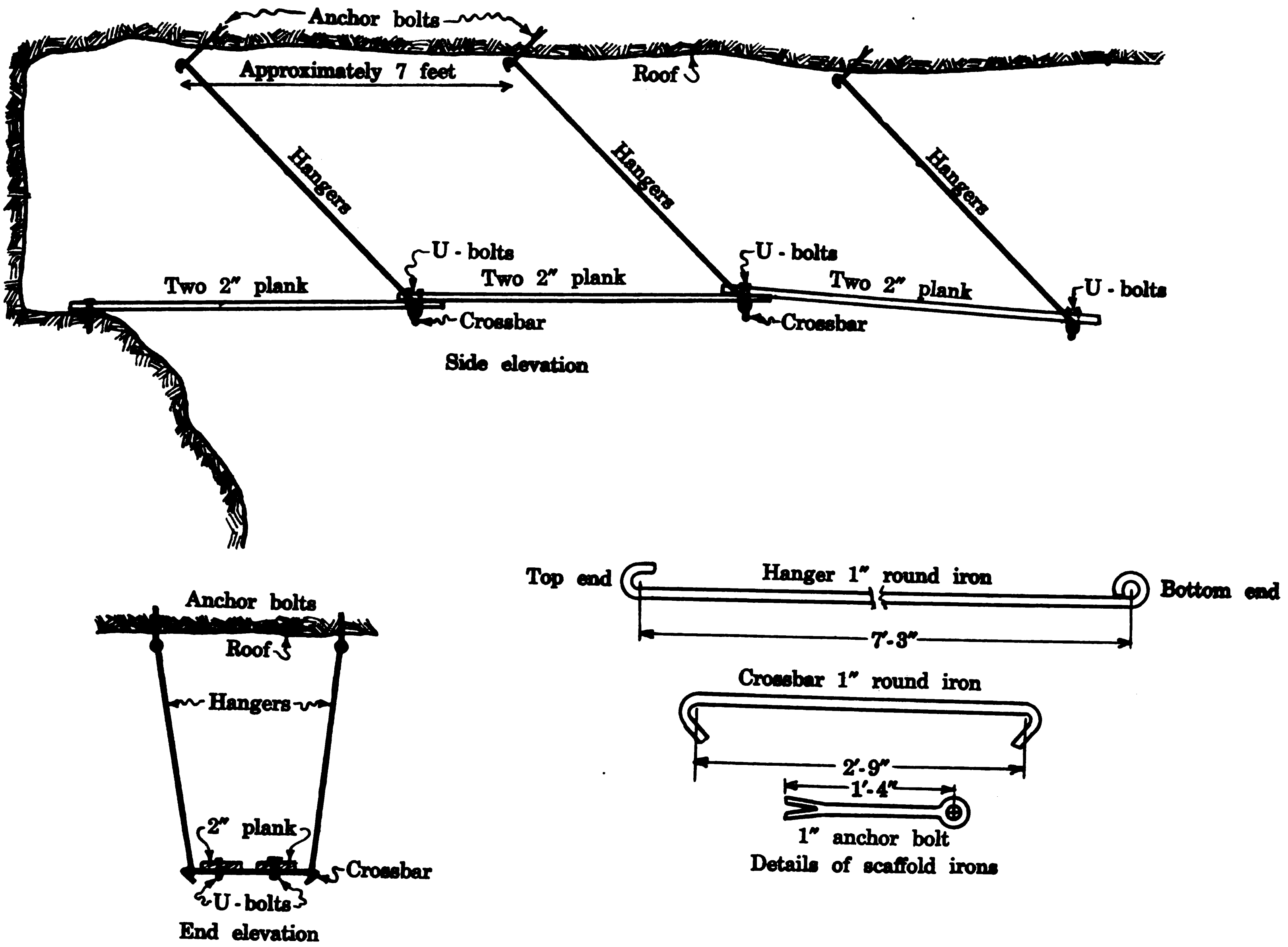

Figure 13. – Suspended scaffold construction

The trolley wire is carried on one side about 12 inches outside the rail and is suspended from substantial iron hanger rods inserted in holes drilled in the roof. Where the roadway passes through high stopes, as it frequently does, galvanized-iron wire cross suspension is employed. Rails are bonded with standard flexible rail bonds arc-welded to the rails.

The main-line haulage locomotives are the manufacturer's standard 13-ton design modified to conform with the mining company's specifications and actually weight about 15 tons. They are driven by two 90 hp. motors and geared to a speed of 15 miles per hour. They pull a maximum of thirty 2½-ton cars; 16-car trains are common, and on one haul where the grade is 2-3/4 per cent against the loads for 4,800 feet 14 cars to the train are customary. This adverse grade is in a long crosscut connecting with a section of the property formerly operated as a separate mine. The average main line haul is 0.79 mile as figured on the ton-mile basis.

Tributary locomotive haulageways are laid with 30-pound rails and mule haulage and hand tramming roads with 30-pound or 20-pound rails. Sidings or loops, as they are termed locally, are established at points near the working faces and at tops and bottoms of inclines for storage of empty and loaded cars. Electric signal systems between these loops and the shaft are employed to notify the locomotive engineer when a loaded train is ready at any particular loop. Mules are usually employed to transfer cars between the loops and loading places. Few mule drivers are employed as such, the loader usually doing his own tramming. Some hand tramming is done on upper levels for tramming short distances to chutes and in some instances in development work.

The standard car is of 2½ tons capacity and has a solid, rectangular body without doors. The car body is of 3/8-inch iron, reinforced by corner angle irons and a heavy band around the top built upon four 5-inch "H" beams. The box is 6 feet 4 inches long by 4 feet wide by 1 foot 9 inches deep; and the top is 3 feet 4 inches above the rail, which is a convenient height for loading with the mechanical shovels and not too high for hand loading. The running gear is of standard railroad type, 16-inch wheels pressed on a 2-15/16-inch axle between railroad-type journal boxes having half brasses and secured to the "H" beams of the car body. Swivel couplings for dumping in rotary dumps without uncoupling are connected to heavy spring drawbars; flat single links with round pins for coupling cars are used.

The cars are hauled to the shaft in trains of 14 to 30 cars on each of the two main levels and are dumped 3 cars at a time without uncoupling by motor-operated rotary dumps. Heavy flat-steel plates under the outside "H" beams of the car and projecting slightly beyond the sides thereof are caught under angle irons running the full length of the rotary dump, thus holding the cars in position on the rails during dumping. The pocket into which the cars are dumped extends from below the lower level station to the upper level and has a total capacity of 1000 tons of ore. The flow of ore from the pocket into the steel manganese-lined chute is controlled by arc gates, operated by hydraulic cylinders. At the lower end of the chute a vertical undercut gate, also operated hydraulically, controls the loading of 2-3/4-ton capacity skips. The space between the upper arc gates and lower undercut gate is not a measuring chute in the true sense of the word, inasmuch as it holds practically 2 skips of rock, but it provides a safety feature in loading. The lower undercut gate is not actuated when the upper arc gates are up and in this way prevents a runaway of ore from the pocket into the shaft.

Hand tramming on the upper levels is done with cars of 1-ton capacity. They are square-body cars without doors and are dumped in a cradle. (Fig. 14.) In this cradle the car is turned end over end. Some of the ore from chutes is wet and tends to pack and build up in the corners and bottoms of the haulage cars. In such instances sheets of heavy oiled paper are employed to cover the bottoms of the cars before they are loaded and have been found to eliminate the trouble. There are only a few chutes in the mine; these are 3 feet in width and are equipped with heavy arc gates operated by long lever arms.

Power enters the mine at 440 volts a.c. and motor-generator sets are employed to transform to 275 volts d.c. for haulage. On long haulageways, 300,000 c.m. feeder lines are carried in parallel with the trolley to hold up the voltage; 440-volt lines are carried to the stopes, where banks of transformers are employed to step down to 220 volts for mechanical shovel operation. A set of semiportable transformers is supplied with each shovel; these are moved from place to place as the shovel moves and are set on the ground a short distance back of the shovel.

At No. 3 shaft, which is used for handling men and supplies, safety gates are installed at each level. These are arranged to cut off the current to the cage hoist automatically when in the open position, so that it is impossible to move the hoist when a gate is open at any level of the shaft.

WAGE, CONTRACT, AND BONUS SYSTEM

Drilling and blasting and loading are generally paid for under a bonus system, which is based upon an old, pre-war scale that covered various classes of work and conditions and was of course much lower than the present basic scale of wages.

In this district the labor turnover is very small, and many of the miners have worked in these mines for a long term of years. Long familiarity with the old prices which had been established for various characters of work and conditions enables the men to figure their own earnings without explanations from supervisory officials, which is a most desirable feature in any contract or bonus system.

All men are guaranteed the basic daily wage ($5.05 per shift). Contracts re figured at the old unit prices, and the amount in excess of the old daily wage imes the number of shifts work is added to the present basic wage rate times the umber of shifts worked to calculate the total amount paid in each case. Eight hours onstitutes a shift.

Drilling. Drillers are paid a set price per foot of advance in development ork and in stopes and stope headings for the number of tons broken, as measured y the number of 2½-ton cars loaded, recorded by car checkers employed for that purose. Drillers furnish only their labor and powder, which is sold to them by the ompany at a fixed price somewhat lower than the actual cost price, while the comany furnishes drilling equipment, tools, and compressed air.

Prices for drifting, raising, and sinking vary with the character of the round, size of excavation, and accessibility of the working place. The machine en carry their own steel to and from the shaft, no "nippers" being employed.

In stopes the unit price varies principally with the height of the stope hough character of ground, accessibility of the stope, etc., may be given considertion also. This price is 16 to 18 cents per ton in low ore and 12 to 14 cents in omewhat higher ore, to a minimum of 8 cents per ton in high stopes or "bluff minng".

Loading. Prices for loading ore in stopes, whether by hand or mechanical oader, vary with the height of stope, tramming distance, and character of the work. Hand loaders tram their own cars and ordinarily use a mule for transferring cars beween the loop and the muck pile. Ordinarily the mechanical shovel operators tram heir own cars in the same manner, unless there is enough muck to warrant mule rivers being employed.

The usual task or "score," as it is termed for a hand mucker, is 20 tons, or loading which he receives only the regular daily wage. In excess of this he eceives a bonus for each extra ton loaded. In places where there is a long tram, and tramming, or badly scattered muck, the score may be reduced to as little as 14 r 16 tons.

In development work it is customary to pay 8 to 10 or 12 hours for mucking ut the round, depending upon the size of the excavation and loading conditions. here the muck is all in one pile, as at the foot of a raise, and other conditions re normal only 8 hours are usually paid, while in driving a slope where loading is ot so easy as in the drift, and where the muckers have to go up and down the slope o operate the hoist 12 hours would be the average pay.

SAFETY WORK

The safety organization of the company, which operates a number of mines and plants, is under the direction of one man, who is responsible only to the general manager. He has a number of assistants, each of whom is assigned the inspection of a definite part of the company's operations.

A continuous campaign in safety education is directed toward developing "safety-mindedness" among the men by means of talks and posters. At the particular mine covered by this report there is a screen at the foot of both entrances to the mine upon which is flashed at the beginning of each shift a safety picture or motto so that it is seen by all the men upon going to work. The picture is changed every day or so, and the apparatus is connected to a time clock so that the light comes on automatically at the right hour and remains on for half an hour.

The general superintendent, superintendent, mine captain, foremen, and bosses together with the safety engineer assigned to each mine, act as the safety committee of that mine. Suggestions from the men as to safety measures are encouraged, but there is no workmen's safety committee as such. The safety score of each foreman and boss is kept carefully, and each boss is responsible for the safety of his men. One of these has a record of approximately 32,000 shifts without a lost-time accident.

The company gives no first-aid training, but first-aid instruction is given periodically by the U. S. Bureau of Mines to selected men, when the Bureau of Mines car is in the district. Due to the fact that the company hospital is close at hand and the attendance of a physician can always be had in a very few moments, first-aid treatment is not encouraged. There are, however a number of trained first-aid men in various sections of the mine to act in cases of extraordinary emergency.

Cost Data for an Average Month - 1929.

Tons ore loaded - mining	49,700	(25,500 loaded mechanically
Tons ore loaded - development	4,300	(24,200 hand loaded
Total tons ore loaded	54,000	
Tons rock loaded	2,000	

	Development[1]	Mining	Total[2]
A. Labor-man hours per ton.			
Breaking (drilling and blasting)	.511	.1625	.208
3/ Mucking (hand)	.687	.416	
3/ Mucking (mechanical)		.061	
3/ Mucking (total)			.231
4/ Transportation (haulage and hoisting) ...			.184
Roofmen		.034	.031
Prospecting			.015
General, includes drainage, machinists, etc.			.126
Supervision			.032
Total			.827
Tons per man shift, all underground labor ..			9.67
Labor, per cent total U. G. Cost	50%	75%	72%
B. Power and supplies.			
Power, kw.h. per ton			
Air compression		3.7	
Hoisting		0.5	
Pumping		5.0	
Mechanical loading		0.2	
Haulage		0.7	
Lighting		0.2	
Total power, kw.h. per ton			10.3
Explosives, 40 per cent ammonia gelatin, pounds per ton	1.324	.459	.55
Power and supplies, per cent of total underground cost	50	25	28
C. Per cent of total cost	18.7	81.3	100

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1. Based on tons of ore and rock loaded from development.
2. Based on total tons ore loaded.
3. Includes mule haulage by loaders between loops and face.
4. Includes operation, maintenance of track and cars, roadmen, stablemen, sand drying, etc.

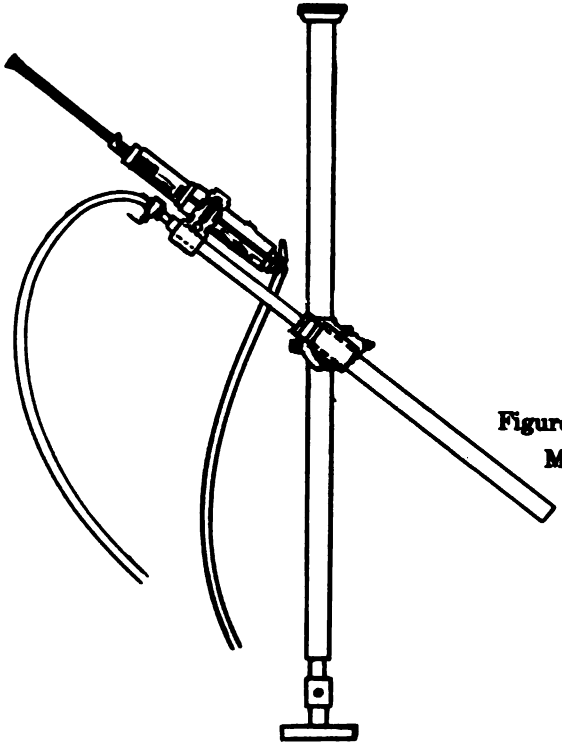

Figure 5. – Mounting for pneumatic feed

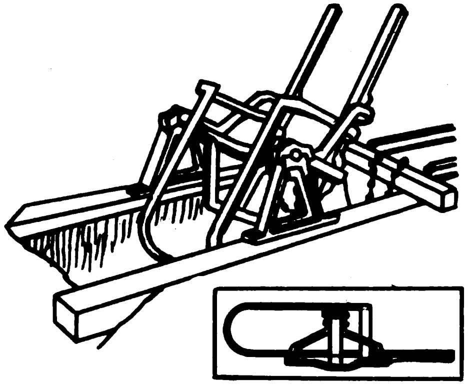

Figure 14. – Cradle dump for 1-ton car from "Fifty Kinks in Metal Mining", 1920, Engineering and Mining Journal

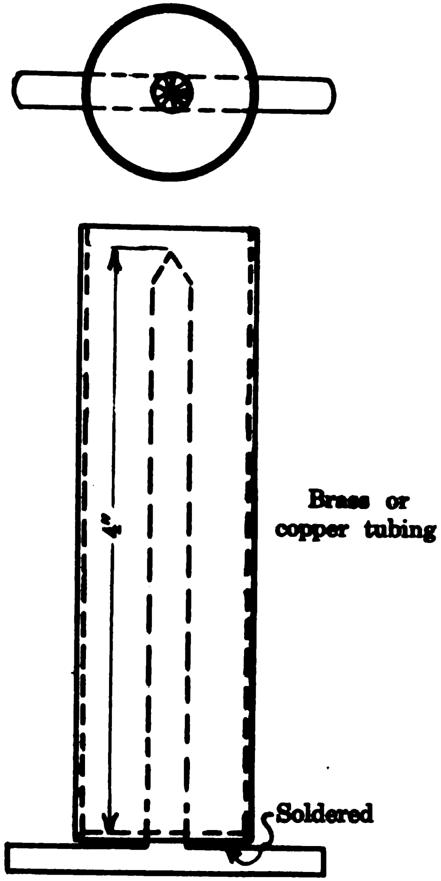

Figure 8. – Powder punch